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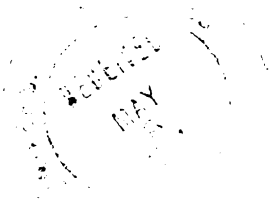

MILLING METHODS AT THE HUGHESVILLE CONCENTRATOR OF THE ST. JOSEPH LEAD CO., HUGHESVILLE, MONT.

BY

WM. O. VANDERBURG

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MILLING METHODS AT THE HUGHESVILLE CONCENTRATOR OF THE ST. JOSEPH LEAD CO., HUGHESVILLE, MONT.[1]

By Wm. O. Vanderburg[2]

INTRODUCTION

This paper describing the milling methods at the Hughesville concentrator of the St. Joseph Lead Co., Hughesville, Mont., is one of a series of similar papers being prepared by the U. S. Bureau of Mines on practices at the various mills in the United States.

The writer expresses his thanks for the courtesies extended by the officials of the company while visiting the property and acknowledges the assistance and helpful suggestions given by R. H. Willcomb, general manager, and Carl Martin, mill superintendent, of the Hughesville Division of the St. Joseph Lead Co.; and E. D. Gardner, supervising engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz., in gathering and compiling the data herein presented.

LOCATION

The concentrator is located in Barker Canyon of the Little Belt Mountains, 66 miles southeast of Great Falls, Mont. The mill site is on a hillside about 2 miles from the mine. Ore is transported from the mine to the upper end of the mill by an aerial tramway 10,250 feet long. The upper terminal of the tramway is 5,930 feet and the lower terminal 5,548 feet above sea level. Factors influencing the choice of the mill site were the availability of ground for tailings disposal and the desirability of maintaining a railroad grade under 3 per cent to the mill. A company-owned standard-gage railroad 10.8 miles long connects the mill with the Neihart Branch of the Great Northern Railroad at Monarch.

The mine is in Judith Basin County and the mill in Cascade County, Mont.

GENERAL

The mill has a daily capacity of 400 tons. An average of 25 men is required to operate the mill on three shifts of eight hours each. Two classes of concentrates are made: A lead concentrate which for the year 1929 averaged 61 per cent of lead, 5.7 per cent of zinc, 50 ounces of silver, and 0.05 ounce of gold; a zinc concentrate averaging 51 per cent of zinc, 1 per cent of lead, 31 ounces of silver, and 0.02 ounce of gold.

The ore occurs in a fissure vein in syenite, and it is mined by the horizontal cut-and-fill system. About 35 per cent of the material blasted is rejected underground as

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waste and is used to fill the mined-out portions of the vein. Selective mining and hand-sorting are practiced to a great extent to obtain a higher-grade product for concentration. Hand-sorting is employed both underground and on the surface. About 8 per cent of the run-of-mine ore is rejected as waste by hand-sorting at the grizzly ahead of the primary breaker.

The water supply for the mill is derived from two sources: Dry Fork Belt Creek, and one of its tributaries, known as Gold Run Creek. All the mill water supply during the summer months has so far been taken from Gold Run Creek. During the winter the tributary creek freezes over and the water gets low, so the mill is supplied with water from Dry Fork Belt Creek. The surge tank for the water supply has a capacity of 85,000 gallons. Water flows by gravity from the dam located on Gold Run Creek through 6,000 feet of 5-inch redwood stave pipe. The water from Dry Fork Belt Creek is elevated to the surge tank in one lift by a single-stage centrifugal pump which has a capacity of 350 gallons per minute. The pipe line consists of 2,500 feet of 4-inch and 800 feet of 6-inch standard iron pipe, the latter connecting the surge tank with the mill. By arrangement of valves which control the flow of water into the mill circuit, the 6-inch portion of the line serves the double purpose of a discharge into the surge tank or into the mill circuit, as required. When the mill is using less water than is being pumped the excess water is forced up the 6-inch line to the surge tank; when more water is required the additional water flows from the surge tank through the 6-inch line to the mill. A check valve is located at the end of the 4-inch line which prevents the water from flowing from the surge tank back through the pump. This is necessary because of a small by-pass at the pump which drains it and the 4-inch line as soon as the pump stops. The by-pass drain prevents freezing of the pump during cold weather in the event that the pump stops. The water requirements of the concentrator amount to about 320 gallons per minute. The mill water is not reclaimed.

Electric power is purchased from the Montana Power Co. The power is generated at hydroelectric plants located on the Missouri River, a short distance from Great Falls. It is transmitted to the property at 23,000 volts, and it is stepped down to 440 volts for use in the mill motors of 3-hp. or larger. The small motors use a voltage of 110 or 220.

ORE TREATED

The economic metals that occur in the ore are lead, zinc, silver, and gold. The average grade of the ore treated for the year 1929 was as follows: 6.21 per cent of lead, 5.01 per cent of zinc, and 9.09 ounces of silver per ton. The ore is practically all sulphide in character. The ore minerals are galena, sphalerite, marmatite (ferriferous sphalerite), and pyrite. A small amount of copper occurs in the concentrates as cupriferous pyrite and chalcopyrite. The galena and sphalerite carry most of the silver. The gangue consists principally of altered syenite and altered rhyolite with subordinate amounts of calcite, barite, quartz, rhodocrosite, and marcasite. The moisture content of the ore as delivered to the concentrator averages about 2.5 per cent.

The ore crushes readily. Fine grinding is necessary to liberate the zinc minerals which are more or less interlocked with the pyrite. The galena is coarsely crystalline, and the classifier return to the rod mill shows the greater part of the galena to be free from gangue.

HISTORY OF CONCENTRATOR OPERATIONS

The first concentrator in the district of 75 tons daily capacity was built in 1911,

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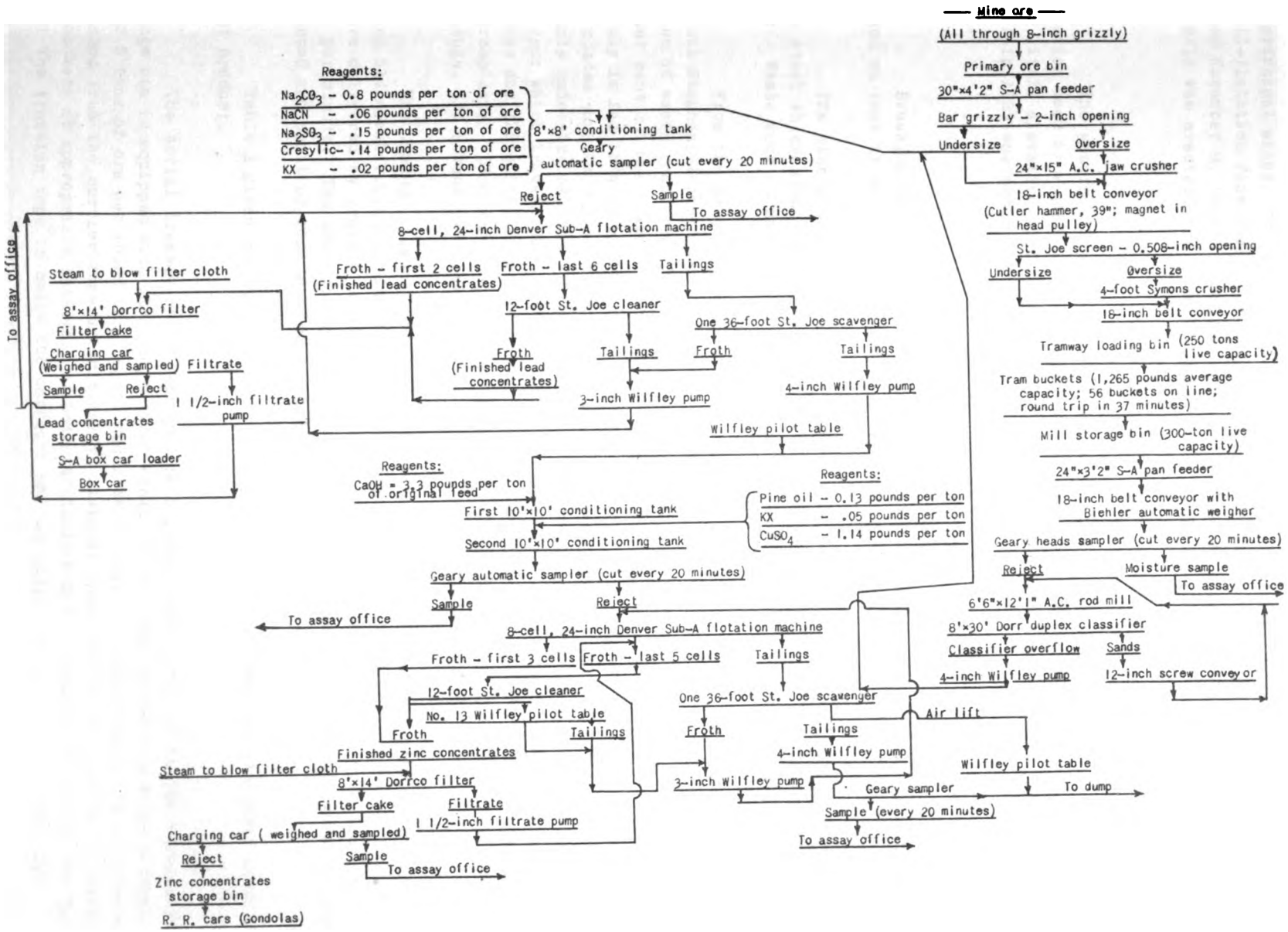

Figure 1.- Flow sheet of crushing plant and mill

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and used gravity methods of concentration. In 1911 this mill was dismantled, and in 1920 it was rebuilt and run intermittently until 1927 when the present company acquired control of the principal mines. The present mill of 400 tons daily capacity was designed and built with an all-flotation flow sheet. Building of the mill was started on May 17, 1928, and was completed November 9, 1928. With the exception of several minor mechanical changes, made after the mill was erected, the flow sheet is essentially the same as first designed.

PRESENT METHOD OF CONCENTRATION

The general flow sheet of the concentrator is shown in Figure 1. The method of flotation used is a selective separation of the lead and zinc sulphides in the original pulp. Very little elevating of the ore or pulp is necessary. The average quantity treated per 24 hours in 1929 was about 312 tons.

Breaking and Crushing

Breaking and crushing of the ore is done at the mine. All the ore is broken underground so that it will pass through an 8-inch grizzly.

The mine ore is delivered into a two-compartment bin with inclined bottoms lined with steel which slope two ways. The capacity of each side of the bin is approximately 50 tons. Waste rock is dumped into one side of the bin and ore into the other.

From the bin the ore is delivered to a bar grizzly with 2-inch spaces, by a 30 by 50 inch Stephens-Adamson pan feeder driven by a 3-hp. motor. At the feeder a sorter removes pieces of waste rock from the ore and drops the waste into a bin at his side. Approximately 30 per cent of the grizzly feed passes through as undersize. The oversize product of the grizzly is fed to a 15 by 24 inch Blake-type crusher. The crusher has manganese steel wearing plates and is driven by a 40-hp. motor operated at 720 revolutions per minute. The grizzly undersize and the Blake-crusher product are delivered by gravity to an 18-inch belt conveyor which travels 300 feet per minute. A 39-inch Cutler-Hammer magnet, operating at 17 amperes and 125 volts, is suspended on a head pulley over the conveyor belt to remove pieces of tramp iron. The conveyor belt discharges on a St. Joe vibrating screen with 0.508-inch openings, the screen oversize feeding a 4-foot Symons cone crusher.

The Symons crusher is direct connected to a 100-hp. alternating-current motor making 514 revolutions per minute. The screen undersize and the product from the cone crusher are delivered by gravity to a rubber-surfaced belt conveyor 18 inches wide traveling 301 feet per minute. The belt conveyor discharges into the tramway loading bin which is flat-bottomed and built of wood and has a capacity of 250 tons, "live" load.

Table 1 gives screen sizes of the Symons-cone-crusher feed and the final crushing-plant product.

The aerial tramway which conveys the crushing-plant product to the concentrator storage bin is equipped with stationary track cable, supporting carriers, and has a capacity of 32.5 tons of ore per round trip, which requires 37 minutes. The buckets are attached and detached from the carrier rope automatically, although supervision is required for spacing the buckets at appropriate intervals at both terminals and for loading and dumping the buckets. The traction rope is below the carriers and the grips are of the underhung type.

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The mill storage bin has a "live" capacity of 300 tons. Ore is fed from this bin to the grinding mill at the rate of 13 tons per hour by a 24 by 38-inch Stephens-Adamson pan feeder. The feeder discharges onto a rubber surfaced conveyor belt inclined at an angle of 10° and operated at a speed of 134 feet per minute. The ore in traveling on this belt is weighed by a Biehler automatic weigher which registers in units of 100 pounds.

Grinding

The conveyor belt delivers the ore to an Allis-Chalmers rod mill 6 feet 6 inches in diameter and 12 feet 1 inch in length, operated at a speed of 18 revolutions per minute by a 240-hp. alternating-current motor making 1,200 revolutions per minute. The gear reduction is 67 to 1. The rod mill is operated in closed circuit with an 8 by 30 foot Dorr duplex classifier, making 21 strokes per minute. The amount of circulating load is about 300 per cent. Manganese steel liners are used in the rod mill, the liner consumption amounting to 0.247 pound per ton of ore. Rod consumption amounts to 1.986 pounds per ton of ore.

The scoop feeder on the rod mill has a stellite lip about 1/8 inch thick to minimize wear. The stellite is welded to the lip locally by means of an acetylene welding torch.

The classifier overflow which comprises the flotation feed contains 23 per cent of solids and is pumped by a 4-inch Wilfley pump to an 8 by 8 foot wooden conditioning tank equipped with a Denver agitating mechanism.

Table 1 gives screen sizes of rod-mill feed and discharge products, and classifier sand and overflow products.

Flotation Lead Circuit

From the lead-circuit conditioning tank the pulp goes to an 8-cell, 24-inch, Fahrenwald Denver Sub-A flotation machine with the cells arranged in series. The first two cells make finished lead concentrates and the other six cells make middlings and tailings. The middlings go to a 12-foot St. Joe flotation machine operating as a cleaner unit and the tailings to a 36-foot St. Joe machine used as a scavenger. The concentrate from the cleaner is a finished lead product. The tailings from the cleaner are returned to the Fahrenwald cells by-passing the lead-circuit conditioning tank. The froth from the scavenger unit is also returned to the Fahrenwald cells. The tailings from the scavenger cells go to the conditioning tanks at the head of the zinc circuit.

Each of the Fahrenwald cells is equipped with a 5-hp. alternating-current motor connected to the impeller shaft by a Texrope drive and making 1,150 revolutions per minute.

The St. Joe flotation cells operate on the bubble column principle. The air necessary to form the bubble column in the 36-foot scavenger and 12-foot cleaner is supplied by a centrifugal blower direct connected to a 27-hp. alternating-current motor making 3,500 revolutions per minute. The capacity of the blower is 4,200 cubic feet of air per minute under a pressure of 0.8 pound per square inch. A decided advantage in the use of the scavenger machine is its ability to function as a safety valve in the event that something goes wrong in the circuit ahead of the machine. The scavenger machine will continue to discharge a clean tailings product, even if it is heavily overloaded for 20 minutes.

A peculiar condition noted in the wear of flotation equipment is that the impellers

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of the lead-circuit flotation cells, the impellers of the conditioning tank, and the aerating pipes in the scavenger cells wear out much faster than similar equipment of the zinc circuit. The impellers of the lead flotation cells handled 110,350 tons of ore while the impellers of the zinc flotation cells up to June 1, 1930, had handled 160,500 tons and appear to be good for about 40,000 tons more.

The selective problem presented in the flotation treatment of this ore is threefold; the separation of the lead and zinc minerals, the recovery of the cupriferous pyrite with its accompanying silver, and the rejection of the marcasite and most of the pyrite which is low in silver.

The amounts and kind of reagents added to the pulp in the conditioning tank at the head of the lead circuit are as follows:

Reagent	Pounds per ton of ore treated
Soda ash	1.466
Sodium cyanide	0.0616
Sodium sulphite	0.1818
Cresylic acid	0.1826
Potassium xanthate	0.0246

No reagents are added to the lead circuit in addition to those introduced in the conditioner as the amount of reagents added to the pulp in the conditioning tank is sufficient to complete the cleaning operation.

Soda ash is added as the alkaline reagent. It improves the froth and tends to make cleaner concentrates.

Sodium cyanide is used as a depressant for the marcasite and pyrite.

Sodium sulphite is used as a zinc depressant. An excess of this reagent in the circuit has no appreciable effect on the appearance of the froth or on the recovery of the lead.

Cresylic acid is used as a frothing agent. The appearance of the froth changes with the lead content of the heads, and the amount of cresylic acid is increased or decreased to take care of this variation.

Potassium xanthate is used principally as a collector.

When the mill was first started zinc sulphate and sodium cyanide were used to depress the zinc in the lead circuit. The change from zinc sulphate to sodium sulphite resulted in lowering the zinc content of the lead concentrates from 6-1/2 to 7 per cent down to 4 to 4-1/2 per cent. Lead concentrates containing about 62 per cent of lead have been found to yield the maximum economic recovery of silver in the lead concentrates.

Table 1 gives the screen sizes of heads to the lead flotation circuit and also screen sizes of the lead concentrates. Table 2 gives a screen-assay analysis of feed to the flotation lead circuit. Table 3 gives the lead, zinc, and iron contents of the individual rougher-cell concentrates, the cleaner-unit products, and the scavenger-machine feed and tailing products of the lead circuit.

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Flotation Zinc Circuit

The tailings product from the lead scavenger unit which is the feed to the zinc flotation circuit goes to two wooden conditioning tanks operating in series, each 10 feet in diameter, 10 feet high, and each equipped with a Denver agitating device. In the first conditioner lime is added, and in the second pine oil, potassium xanthate, and copper sulphate.

The flotation cells of the zinc circuit are identical with those of the lead circuit.

From the second conditioning tank the pulp passes to a Fahrenwald machine with 8 cells arranged in series. These cells make three products: Finished concentrates from the first three cells, middlings from the last five cells, and tailings. The middlings froth goes to a 12-foot St. Joe cleaner which makes finished concentrates and tailings. The tailings are returned to the first Fahrenwald cell. The tailings from the Fahrenwald cells go to a 36-foot St. Joe scavenger machine which makes a middlings froth which is returned to the head of the Fahrenwald cells and tailings which go to waste.

The only dilution of the pulp in the zinc circuit is from the water added to the concentrates launders and returned to the circuit by the filtrate pump. Diluting water is kept as low as possible. The amount and kind of reagents added to the pulp in the conditioning tanks ahead of the zinc circuit are as follows:

Reagent	Pounds per ton of ore treated
First conditioning tank:	
Lime	3.337
Second conditioning tank:	
Pine oil	0.125
Potassium xanthate	0.0399
Copper sulphate	1.210

Lime is used as a depressant for the marcasite and pyrite.
Pine oil is used as the frothing agent.

Potassium xanthate is used as a froth stiffener and as a collector. In the zinc circuit the xanthate is very sensitive; a deficiency increases the zinc content of the tailings and an excess produces concentrates of high iron content. An excess of pine oil will also increase the amount of iron in the concentrates. An increase in the amount of xanthate added necessitates an increase in the amount of pine oil.

Copper sulphate is the reactivating agent for the zinc minerals which have been temporarily depressed in the lead circuit.

Table 1 gives screen sizes of the zinc-flotation concentrates and the final concentrator tailings.

Table 2 gives a screen-assay analysis of the final concentrator tailings.

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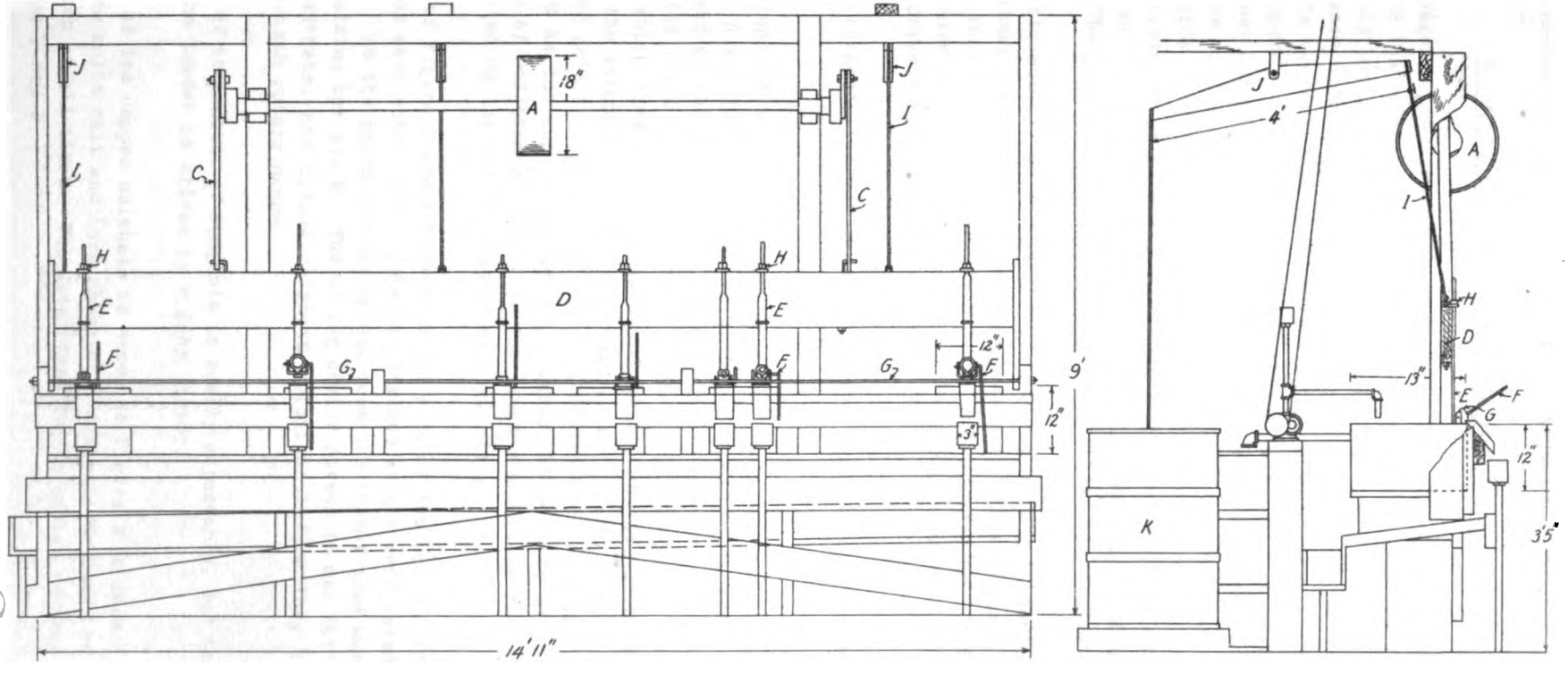

Figure 2.- Reagent feeder

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Table 3 gives the lead, zinc, and iron contents of the individual rougher-cell concentrates, the cleaner-unit products, and the scavenger-machine products of the zinc section.

Conditioning of Pulp and Feeding of Reagents

All the reagents used are added to the conditioning tanks as solutions, except the soda ash added to the lead circuit and the lime added to the zinc circuit. The soda ash and lime are fed uniformly at the rate of 19 and 43-1/2 pounds per hour, respectively, by conveyor-belt feeders each driven by a 1/4-hp. motor. The rate at which the reagents are fed from hoppers onto the conveyor belts is controlled by a cone drive and a gate at the discharge end of the hoppers. All reagents for the flotation of lead, except cyanide, are added to the lead conditioner through a stationary column 12 inches in diameter and 5 feet 4 inches long. The feed to the conditioner also enters through this column. The cyanide is added to the conditioner outside the column in order that the sodium sulphite will have the first action on the sphalerite as cyanide has a tendency to float sphalerite. By omitting the cyanide entirely, lead concentrates assaying about 45 per cent of lead and 2.7 per cent of zinc are produced. The time of conditioning pulp for the lead circuit is 5 minutes.

As previously noted, two conditioning tanks, operating in series, are used in the zinc circuit. With these conditioners in series, sufficient time is given for the conditioning of the pulp, and the use of an excessively large unit is avoided. The conditioning time for the pulp of the zinc circuit is 30 minutes. The impeller in each conditioning tank is driven by a Texrope drive from a 5-hp. motor.

All liquid reagents, except copper sulphate, are fed by the reagent feeder shown in Figures 2 and 3.

The operation and control of this reagent feeder are as follows: Referring to Figure 2, the pulley A on the drive shaft B actuates the rods C, which are eccentrically connected to either end of the drive shaft. The length of the eccentric arms at the ends of the drive shaft and the ratio of the motor and drive shaft pulleys are such that the rods C lift the board D to which they are attached about 10 inches, 16 times per minute. To the board D are fastened the straps E. At the lower end of each strap a small iron cup is pivoted as shown in greater detail in Figure 3. To each cup is secured a small iron rod F which slides upon the rod G as the cup is raised and lowered. The angle of tip of each cup determines the quantity of solution emptied into the launder each time the cup is raised. The angle of tip is adjusted by the nut R attached to each strap.

The rods I of Figure 2 are attached to the ends of the board D and the rocker arms J. To the other end of each rocker arm a rope is attached with a perforated pail tied to the free end of each rope. As the board D moves up and down the rocker arms move the pails which are suspended in the mixing barrels K. The mixing device serves to mix thoroughly the sodium cyanide, potassium xanthate, and sodium sulphite solutions before they are pumped to the reagent feeder by 1/2-inch rotary pumps.

The control of the amount of reagents is easily adjustable, and the device is accurate and reliable. The feeder is driven by a 3-hp. motor.

The feeding of the copper sulphate as shown in Figure 4 is done from a wooden barrel, which contains the solid salt and through which a controlled volume of water flows giving a saturated solution at all times. The water feed to the barrel is controlled by the reagent feeder of Figures 2 and 3.

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This method of feeding copper sulphate has been found satisfactory, and due to the even temperature maintained in the mill by the heating system there is little tendency for the saturated solution to crystallize before reaching the conditioning tank.

All reagents are added to the conditioning tanks from the central floor, thus giving control of the entire plant from one point.

SAMPLING AND MILL CONTROL

Samples of the rod-mill feed, the heads of the lead and zinc circuits, and the mill tailings from the zinc-scavenger unit are taken by Geary samplers operated by an electrically controlled mechanism. The rod-mill sample is taken principally for the determination of moisture in the ore, although it is also used as a check on the heads sample. The sample cuts are taken simultaneously every 20 minutes, by all four samplers. The samples thus taken are combined into four composite samples for each shift and assayed for lead, zinc, and iron. The mill-tailings samples are not assayed for iron.

Pipe samples are taken of the lead and zinc concentrates. After weighing each hopper load of concentrates the weigher takes a pipe sample. A hopper load of lead concentrates weighs about 4,000 pounds, and one of the zinc concentrates weighs about 3,000 pounds. The pipe samples are combined into composite samples for each class of concentrates for each shift. The zinc-concentrates composite samples are assayed for insoluble matter as well as lead, zinc, and iron.

In addition to the samples mentioned the mill operations are controlled by the use of three 18 by 40 inch Wilfley pilot tables, which continuously treat a portion of the tailings from the lead-scavenger cells, the concentrates from the zinc-cleaner cells, and the mill tailings from the zinc-scavenger unit.

Each pilot table is driven by a 1/4-hp. motor. The use of the small concentrating tables as continuous vanning plaques furnishes a continuous visual check on the mill operations.

The standard La Motte comparator is used for the determination of alkalinity - or, more accurately, for the determination of pH values. Thymol blue is used in the lead circuit and bromothymol blue for the mill water. The pH value of the mill water is 7.6, that of the pulp in the conditioner ahead of the lead circuit is 8.4, and the pH value of the tailings pulp is approximately 11.0.

The control of the lead and zinc circuits is not restricted to pH readings but is based largely on the appearances of the froths, and the use of vanning plaques. The flotation operators have become proficient in handling the circuits by visual inspection, and the changing of the amounts of reagents is largely left to their judgment.

FILTERING

Finished concentrates products are produced in the lead circuit by the first two Fahrenwald cells and the St. Joe cleaner unit.

On the zinc side the first three Fahrenwald cells and the St. Joe cleaner unit produce finished concentrates.

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In both the lead and zinc circuits the finished concentrates are sent to a Dorrco filter, 8 feet in diameter and 14 feet long driven by a 5-hp. motor. The material used for the filtering medium is Palma twill, style 31, made by the Filter Fabrics Corporation. Each filter has 336 square feet of filtering area. On the lead filter the average life of the filter cloth is about five weeks and on the zinc side about three weeks. The shorter life of the cloth on the zinc side is due to the fact that the cloth blinds more rapidly on the zinc side.

The filter cake is dewatered by suction before it reaches the discharge point at the top of the drum. A pulsating valve at the top of the drum automatically reverses from suction to compression three times while the cake passes the top section. The pulsations discharge the cake onto a conveyor belt. Constant use of the filter tends to clog the pores in the cloth with slimes. This is overcome by using live steam an average of about an hour each day. The steam is admitted through the blow connections and the blow is cut off at this time. The steam tends to loosen the pores in the filter cloth and to force out the slimes that have collected in them.

The vacuum for each filter, equal to 21 inches of mercury, is supplied by an Ingersoll Rand, 22 by 8 inch, belt-driven vacuum pump. The pressure used for freeing the filter cake from the cloth is 5 pounds per square inch. Each vacuum pump is driven by a 30-hp. motor operating at 1,165 revolutions per minute.

The moisture content of the lead and zinc filter cakes is 7 and 9 per cent, respectively.

The filtrate on the lead side is returned to the head of the Fahrenwald cells, and the filtrate on the zinc side is returned to the fourth Fahrenwald cell.

WEIGHING AND LOADING OF CONCENTRATES

The lead and the zinc concentrates are conveyed by the traveling belt inside of each filter drum to the weighing hoppers. Each hopper is connected to Howe scales for weighing the concentrates before shipment. Each hopper is equipped with a 1/4-inch sheet-iron partition as shown in Figure 5 so that when the hopper is dumped at either side, the concentrates fall by gravity to either of two bins for each class of concentrates. Each bin compartment has an inclined bottom lined with sheet iron, and will hold 105,000 pounds of zinc concentrates or 88,000 pounds of lead concentrates.

The zinc concentrates are shipped in bottom-dump, steel railroad cars, and the lead concentrates in box cars. The box cars are used so that they may be utilized for a return haul with mill supplies. The box cars are loaded by means of a Stephens-Adamson box-car loader. With this loader the lead concentrates are run by gravity into the hopper on the machine and are thrown tangentially off an endless belt driven by a small motor and Texrope drive. The carrying surface of the belt travels in a concave curve over rollers, and as the load on the belt travels around the curve formed by the belt, the speed of travel throws the material to the end of the car. The discharge angle of the belt can be varied to pile the material to any height. A box car can be loaded with this machine in about two hours.

TAILINGS DISPOSAL

The tailings are pumped to a pond through 1,400 feet of 4-inch standard iron pipe

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by a 4-inch centrifugal pump direct connected to a 30-hp. motor making 1,160 revolutions per minute. The delivery pipe is laid on oil barrels or 2 by 6 inch crossed timbers, along the crest of the tailings pond which is located in a shallow swale. A T-fitting is placed at the junction of each 16-foot length of pipe. In the open end of each T-fitting a 4 by 1 inch reducer is fitted, through which the tailings are diverted into the pond. By plugging the 1-inch holes and by means of valves placed at intervals along the delivery line, the tail ings can be directed to any portion of the pond. The crest of the pond is built up continuously with tailings shoveled by hand. The excess water is drained off through box launders underneath the pond.

LABOR

The mill labor is composed entirely of American-born men, who are efficient and steady. All mill work is done on company account and the wage scale for the principal classifications of employees as of April, 1930, was as follows:

Crusherman	$5.50
Rod-mill operator	5.00
Flotation operator	5.50
Filterman	4.50
Sampler and reagent mixer	5.00
Loader	4.50
Tramway operator	4.50 to 5.00
Tailings-pond man	4.50

TABLES

Table 1 gives screen sizes of Symons cone-crusher feed, concentrator feed, intermediate and final concentrator products. Table 2 gives screen-assay analyses of flotation feed and flotation-tailings products. Table 3 gives the lead, zinc and iron contents of concentrates produced by the individual rougher cells and also the lead, zinc, and iron contents of cleaner and scavenger unit products for both the lead and zinc circuits.

Table 4 shows metallurgical data for the year 1929. Table 5 gives average assays of concentrator heads and final products and the percentage distribution of the silver, lead, and zinc in concentrator final products for the year 1929. Table 6 shows cost summaries in units of labor, power and supplies for the year 1929.

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Table 1.- Screen sizes of crushing-plant and concentrator products

Screen size	Symons cone-crusher feed		Rod-mill feed		Classifier feed		Classifier sands		Classifier overflow		Lead concentrates		Zinc concentrates		Tailings	
	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent
On 3-inch ring	3.6	3.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 2 1/2-inch ring	7.7	11.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 2-inch ring	9.0	20.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 1-inch square	26.2	46.5	0.7	0.7	-	-	-	-	-	-	-	-	-	-	-	-
On 1/2-inch square	-	-	12.1	12.8	-	-	-	-	-	-	-	-	-	-	-	-
On 3 mesh	23.3	69.8	37.2	50.0	-	-	-	-	-	-	-	-	-	-	-	-
On 4 mesh	3.4	73.2	10.1	60.1	-	-	-	-	-	-	-	-	-	-	-	-
On 6 mesh	3.7	76.9	6.8	66.9	-	-	-	-	-	-	-	-	-	-	-	-
On 10 mesh	6.9	83.8	11.1	78.0	-	-	-	-	-	-	-	-	-	-	-	-
On 14 mesh	3.3	87.1	4.7	82.7	-	-	-	-	-	-	-	-	-	-	-	-
On 20 mesh	2.9	90.0	3.8	86.5	0.4	0.4	0.6	0.6	-	-	-	-	-	-	-	-
On 28 mesh	1.9	91.9	2.7	89.2	0.9	1.3	1.1	1.7	-	-	-	-	-	-	-	-
On 35 mesh	2.4	94.3	3.1	92.3	3.1	4.4	4.3	6.0	-	-	-	-	-	-	-	-
On 48 mesh	1.8	96.1	2.3	94.6	10.5	14.9	12.9	18.9	1.1	1.1	-	-	-	-	2.2	2.2
On 65 mesh	1.3	97.4	1.6	96.2	16.1	31.0	17.9	36.8	9.0	10.1	0.2	0.2	0.8	0.8	11.7	13.9
On 80 mesh	0.3	97.7	0.4	96.6	6.8	37.8	7.7	44.5	4.4	14.5	-	-	1.0	1.8	4.9	18.8
On 100 mesh	0.6	98.3	0.8	97.4	14.8	52.6	16.2	60.7	10.3	24.8	0.8	1.0	4.8	6.6	11.2	30.0
On 150 mesh	-	-	-	-	18.1	70.7	20.3	81.0	14.1	38.9	2.9	3.9	14.9	21.5	14.8	44.8
On 200 mesh	-	-	-	-	10.5	81.2	10.2	91.2	13.6	52.5	7.1	11.0	18.7	40.2	13.8	58.6
On 250 mesh	-	-	-	-	3.8	85.0	2.4	93.6	-	-	4.4	15.4	7.6	47.8	-	-
Through 250 mesh	-	-	-	-	15.0	-	6.4	-	-	-	84.6	-	52.2	-	-	-

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Table 2.— Screen-assay analyses of typical flotation feed and tailings

FLOTATION FEED TO LEAD CIRCUIT

Screen size, mesh	Weight, pounds in 100	Analyses, per cent		Weight of metals, pounds		Per cent of total	
		Lead	Zinc	Lead	Zinc	Lead	Zinc
Composite feed	100.0	6.90	3.79	6.899	3.790	100.0	100.0
On 48	1.1	0.79	0.56	0.009	0.006	0.1	0.2
On 65	9.0	1.63	1.12	0.147	0.101	2.1	2.6
On 80	4.4	1.29	1.28	0.057	0.056	0.8	1.4
On 100	10.3	1.60	2.50	0.165	0.258	2.4	6.8
On 150	14.1	1.84	4.75	0.259	0.670	3.8	17.8
On 200	13.6	3.50	5.00	0.476	0.680	6.9	17.9
Through 200	47.5	12.18	4.25	5.786	2.019	83.9	53.3

FLOTATION TAILINGS

Screen size, mesh	Weight, pounds in 100	Analyses, per cent		Weight of metals, pounds		Per cent of total	
		Lead	Zinc	Lead	Zinc	Lead	Zinc
Composite tailings	100.0	0.31	0.34	0.314	0.343	100.0	100.0
On 48	2.2	0.16	0.22	0.004	0.005	1.3	1.5
On 65	11.7	0.27	0.40	0.032	0.047	10.2	13.7
On 80	4.9	0.30	0.48	0.015	0.024	4.8	7.0
On 100	11.2	0.31	0.28	0.035	0.031	11.1	9.1
On 150	14.8	0.27	0.48	0.040	0.071	12.7	20.8
On 200	13.8	0.25	0.30	0.035	0.041	11.2	11.7
Through 200	41.4	0.37	0.30	0.153	0.124	48.7	36.2

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Table 3.—Analyses of flotation products under normal operating conditions

Product	Analyses, per cent		
	Lead	Zinc	Iron
No. 1 lead-cell concentrates	72.8	2.9	3.8
No. 2 lead-cell concentrates	68.2	3.75	5.1
No. 3 lead-cell concentrates	51.0	7.1	10.7
No. 4 lead-cell concentrates	37.6	8.8	14.8
No. 5 lead-cell concentrates	20.6	10.55	21.3
No. 6 lead-cell concentrates	14.0	11.25	24.0
No. 7 lead-cell concentrates	10.2	11.65	24.7
No. 8 lead-cell concentrates	9.0	11.6	24.2
Lead-scavenger feed	0.50	4.2	11.3
Lead-scavenger tailings	0.35	4.05	10.5
Lead-cleaner concentrates	59.8	7.10	7.3
Lead-cleaner tailings	8.08	13.55	18.5
No. 1 zinc-cell concentrates	1.10	50.8	6.0
No. 2 zinc-cell concentrates	1.29	50.8	6.2
No. 3 zinc-cell concentrates	1.47	48.2	7.0
No. 4 zinc-cell concentrates	1.65	36.0	10.0
No. 5 zinc-cell concentrates	1.94	38.8	9.8
No. 6 zinc-cell concentrates	2.77	40.4	9.0
No. 7 zinc-cell concentrates	2.57	40.0	10.8
No. 8 zinc-cell concentrates	2.84	28.1	11.4
Zinc-scavenger feed	0.20	0.7	-
Zinc-scavenger tailings	0.14	0.38	-
Zinc-scavenger concentrates	0.57	3.8	-
Zinc-cleaner concentrates	1.86	50.2	6.8
Zinc-cleaner tailings	2.2	9.4	-

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Table 4.— Metallurgical data for the year 1929

Total ore treated, dry tons	106,537
Moisture in ore to mill, per cent	2.51
Hours operated per day	24
Days operated	341.52
Average ore treated per 24 hours, tons	311.85
Total concentrates produced, dry tons	18,762.22
Average lead concentrates produced per 24 hours, dry tons	30.296
Average zinc concentrates produced per 24 hours, dry tons	24.641
Per cent of total lead in lead concentrates	95.46
Per cent of total lead in zinc concentrates	1.48
Per cent of total lead floated	96.94
Per cent of total zinc in zinc concentrates	80.76
Per cent of total zinc in lead concentrates	11.08
Per cent of total zinc floated	91.84
Per cent of total silver in lead concentrates	53.61
Per cent of total silver in zinc concentrates	26.81
Per cent of total silver floated	80.42
Ratio of concentration, lead section	10.29
Ratio of concentration, zinc section	12.65
Net water consumption per ton of ore, tons	6
Average solids in lead-circuit pulp, per cent	23
Average time of conditioning in lead circuit, minutes	5
Average time of conditioning in zinc circuit, minutes	30
Average temperature of lead-circuit pulp, degrees F	35
Average temperature of zinc-circuit pulp, degrees F	40
Rod consumption per ton of ore, pounds	1.986
Liner consumption per ton of ore, pounds	0.247

Table 5.--Average assays and distributions of silver, lead and zinc in final concentrator products for year 1929

Product	Assays				Per cent of total metals		
	Gold, ounces per ton	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Silver	Lead	Zinc
Heads	-	9.09	6.21	5.01	100.00	100.00	100.00
Lead concentrates	0.049	50.19	61.06	5.71	53.61	95.46	11.08
Zinc concentrates	0.022	30.86	1.17	51.20	26.81	1.48	80.76
Tailings	-	2.15	0.23	0.49	19.58	3.06	8.16

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Table 6.—Summary of costs in units of labor, power and supplies for year 1929

Tons of dry ore treated: 106,537.

Dry tons of concentrate produced: 18,762.22.

Labor (tons per man per 8-hour shift)	
Crushing and sorting	78.789
Grinding	104.141
Flotation	104.141
Filtering	104.141
Weighing and loading	156.212
Sampling and mixing reagents	312.425
Maintenance	156.212
Supervision	312.425
Assaying	156.212
Warehouse	312.425
Tailings pond	156.212
Power (kw.h. per ton of ore treated)	
Crushing	2.682
Grinding	11.174
Flotation	11.514
Filtering	4.111
Miscellaneous	1 609
Total	31.090
Reagents (pounds per ton of original ore treated)	
Sodium carbonate	1.466
Sodium cyanide	0.0616
Sodium sulphite	0.1818
Potassium xanthate	0.0754
Cresylic acid	0.1826
Pine oil	0.1250
Lime	3.337
Copper sulphate	1.210
Miscellaneous	
Rods (pounds per ton)	1.986
Liners (pound per ton)	0.247

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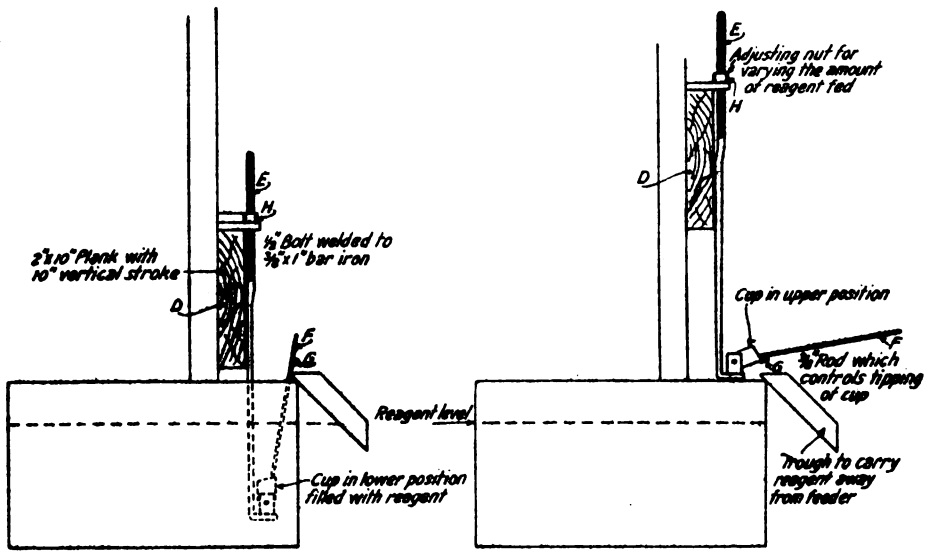

Figure 3.-Details of reagent feeder

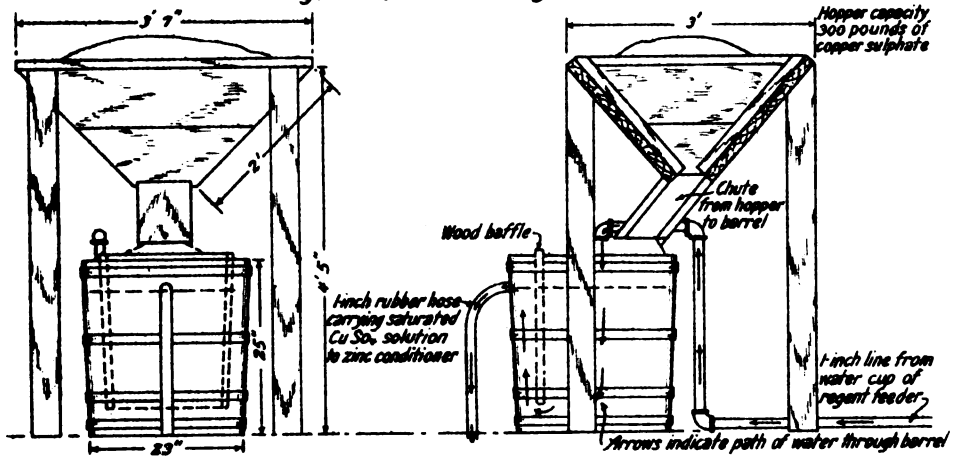

Figure 4.- Copper-sulphate feeder

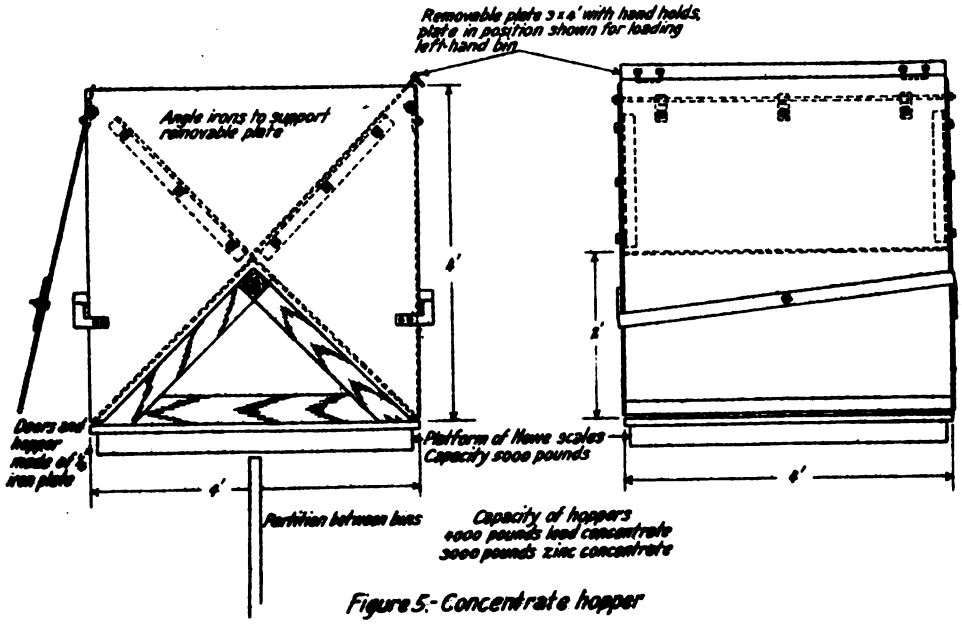

Figure 5.- Concentrate hopper

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